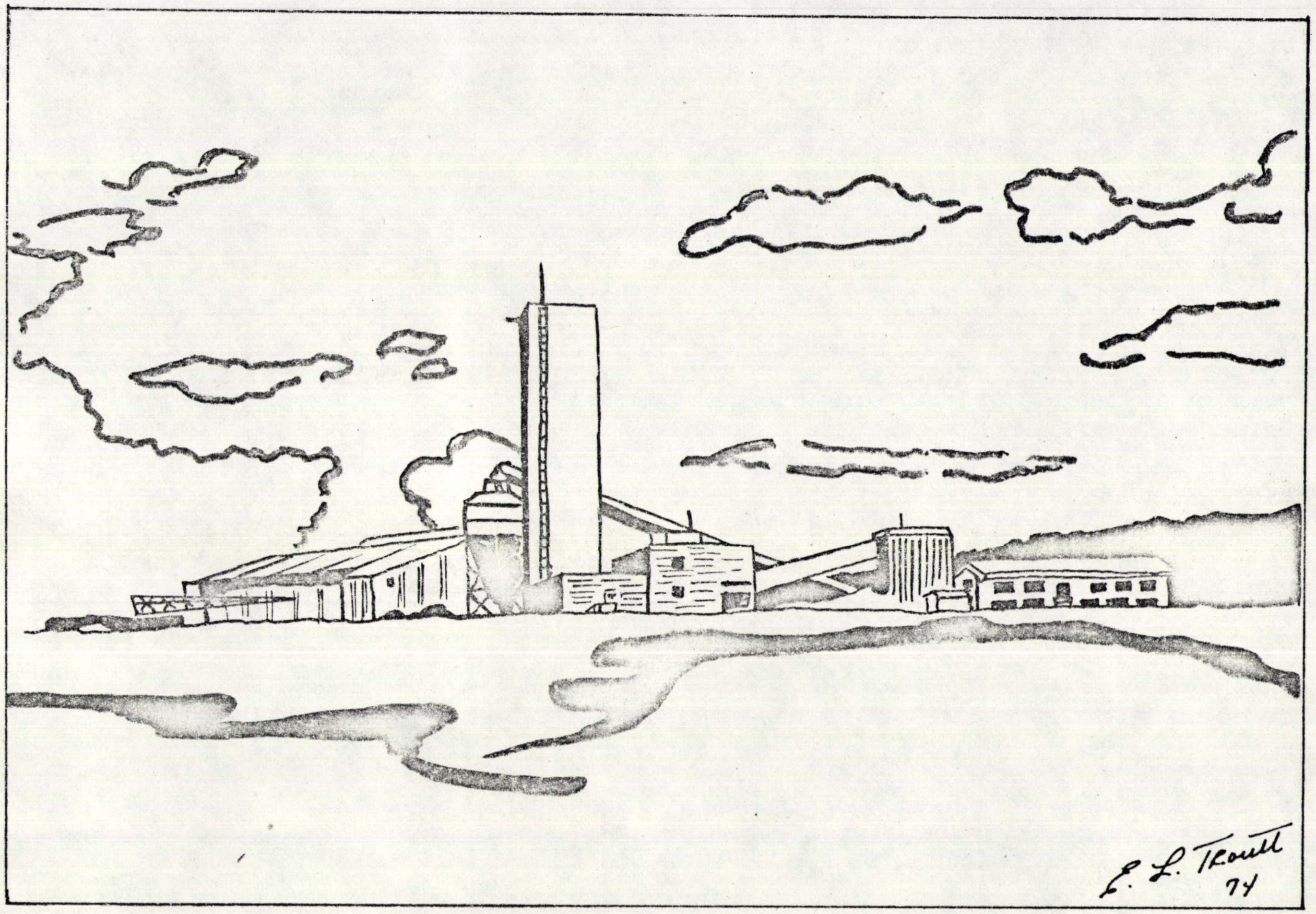


Rio Algom Corporation
Lisbon Mine

File - Rio Algom.
Ray
Jim

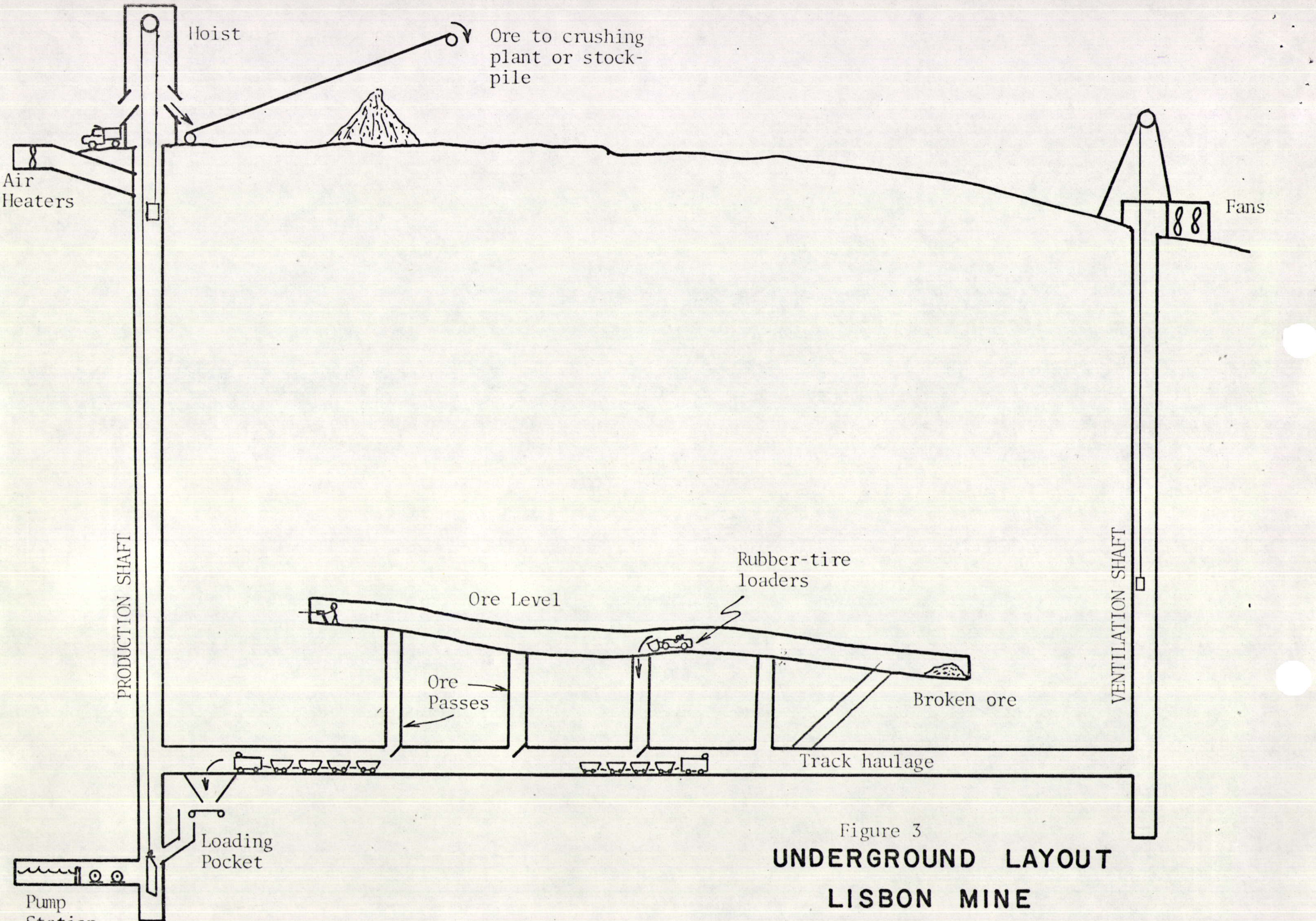


Figure 3

UNDERGROUND LAYOUT

LISBON MINE

RIO ALGOM CORP.

MOAB, UTAH

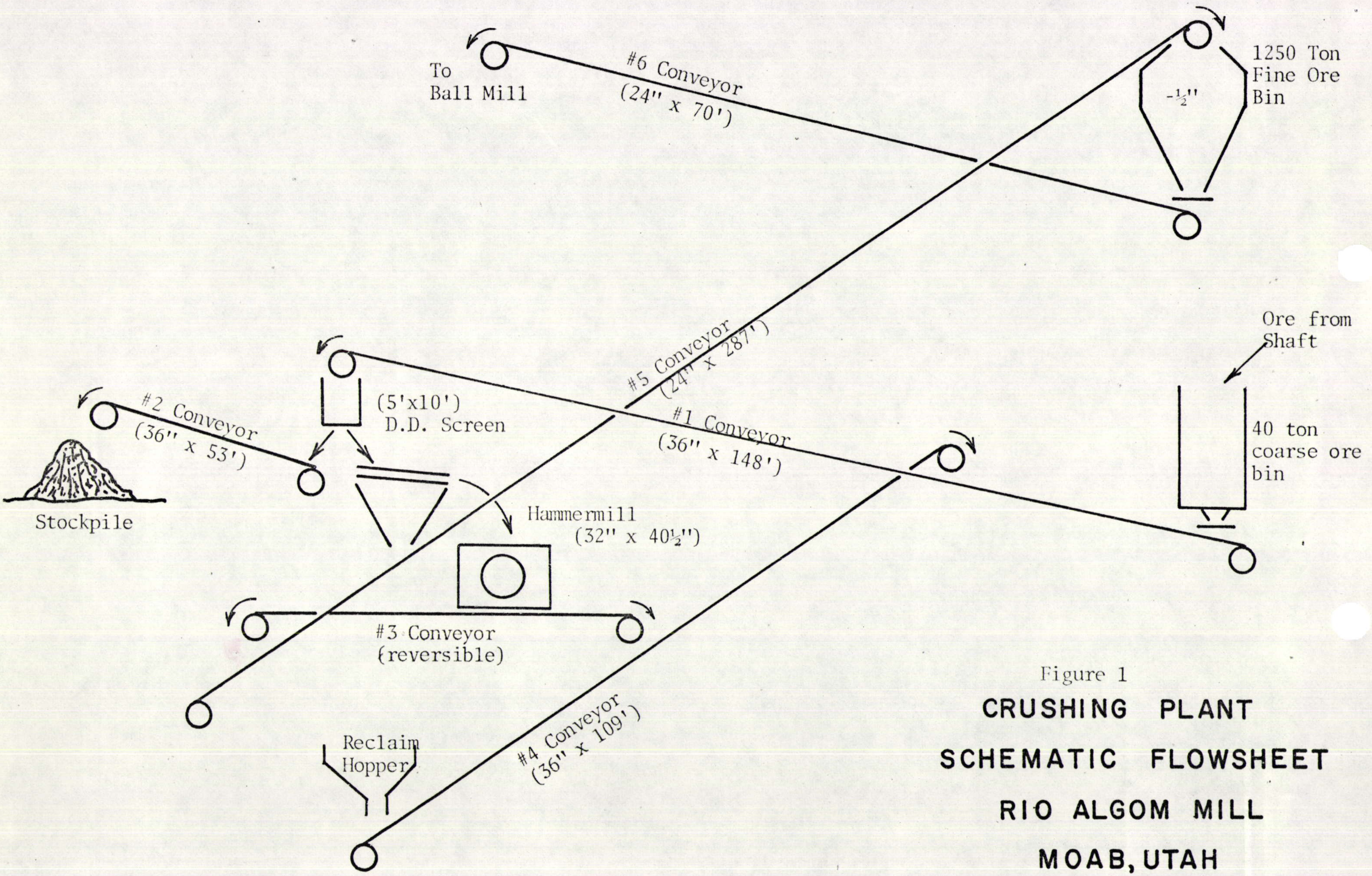


Figure 1

CRUSHING PLANT

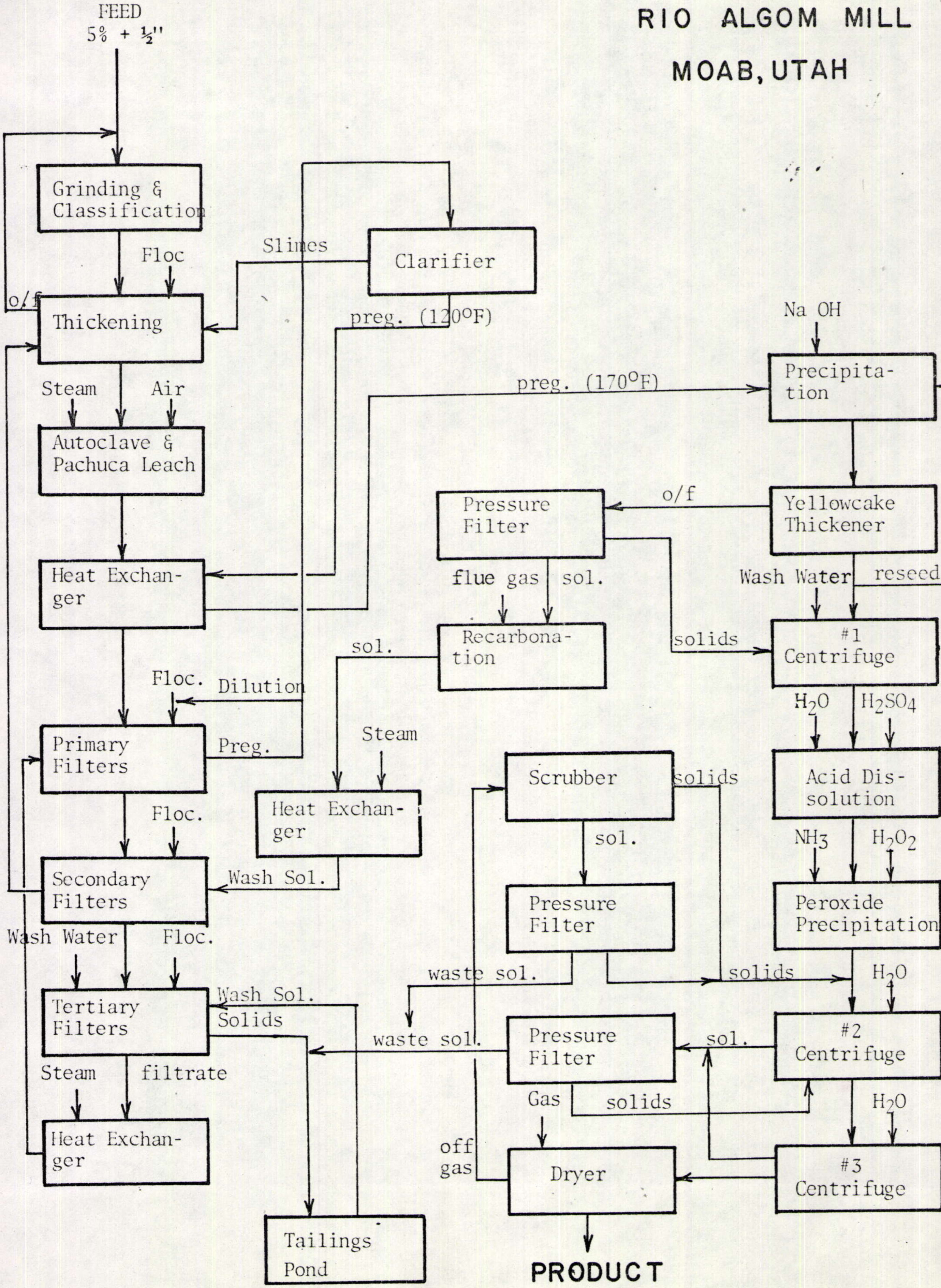
SCHEMATIC FLOWSHEET

RIO ALGOM MILL

MOAB, UTAH

FIGURE 2

SIMPLIFIED MILL FLOWSHEET
RIO ALGOM MILL
MOAB, UTAH



RIO ALGOM CORPORATION
LISBON OPERATIONS

1) INTRODUCTION

The Lisbon Mine is located at an elevation of 6,700 feet in the Lisbon Valley some 35 miles Southeast of Moab, Utah.

The mineralization in the orebody occurs at a depth of approximately 2,550 feet in the Mossback member of the Chinle formation of the Triassic age, within a few feet of the unconformity between the Chinle and the Cutler formation of Permian age. The ore texture grades from conglomerate to sand, to siltstone, to mud. A typical assay showed the following percentages of other elements to be present with the Uranium CO_2 9.70, S .45, V .005, Ca 10.43, and Th .12.

The mine was developed and mill constructed at a capital cost of more than 23 million dollars. The company is processing 600 to 700 tons per day, with the mill working 7 days per week. The mine will deliver a total of 7.9 million pounds of Uranium to Duke Power Company, of Charlotte, North Carolina, by 1980 in accordance with sales contracts. A further 500,000 lbs. will be delivered to a Swedish utility.

Approximately 200 people are employed and most make their homes in the Moab, La Sal, and Monticello areas.

2) MINE DEVELOPMENT (See Fig. 3)

Two 18 foot diameter concrete lined shafts were sunk to depths of 2,665 and 2,686 feet. The former is the production shaft used for fresh air entry, production hoisting and service. A 7½ ton skip in balance with a large cage operates on rope guides in conjunction with a friction hoist mounted on top of the 160 foot high headframe.

Installed at the #2 shaft are ventilating fans capable of exhausting 250,000 cu. ft. of air per minute. An emergency man hoisting facility is also installed here.

Crews driving from both shafts completed the 4300 feet of connecting drift in November, 1971. A four month delay was experienced in the advance from the number 2 shaft when crews drilled into a large water filled fault. Water inflow was greater than pumping capacity, and the drift and shaft were flooded. The area was dewatered by using deep well pumps suspended from the surface. Water not used by the mine from this shaft is diverted to local stock watering ponds for the benefit of ranches in the area.

After the shafts were connected, development by rail haulage continued in waste about 100 feet below the ore horizon. Entries to the orebody were made via inclines with initial service and development muck being handled through these ramps.

Raise boring from the sub-ore haulage to the development ore drives above is being made to provide boxholes and ventilation openings.

The ore is developed on a double entry system which outlines stope areas in roughly 100 foot square blocks. This method allows flexibility in blending various grades to produce an optimum grade of feed to the mill. Pillar drifts are driven through these blocks and pillars are left in place for roof support.

Drilling with conventional rock drills and electric blasting with prilled explosive breaks the ore at the faces.

Some areas contain quantities of perched or connate water and these must be drained by drill holes up from the haulage level. Test holing, probing for water, preceeds all development faces.

Ore is removed from the stoping area by load-haul-dump equipment to the boxholes where diesel locomotives haul the 3½ ton cars to the loading pocket. Automated loading and hoisting of the skip takes the ore to a surface bin from where it is conveyed to crushing facilities or, if necessary, to a surface stockpile.

3) MILLING

The Lisbon Mill is a typical carbonate leach plant. It was originally designed for 500 TPD, but is being expanded to run 750 TPD.

3-1 CRUSHING CIRCUIT AND FINE ORE STORAGE (See Fig. 1)

The Crushing circuit is designed for a crushing rate of 110 tons per hour. It is designed in such a way that it can be operated on either open circuit or closed circuit. If the ore is not desirable as direct feed to the circuit, it can be stockpiled for blending via the conveyor system.

The run of the mine ore is hoisted to a 40 ton storage bin on the surface. From there it is fed, via a pan feeder, into #1 conveyor which takes it to a hopper with flop gate arrangement. If the ore is unacceptable, the flop gate will divert the ore to #2 conveyor which delivers it to the stockpile. If the ore is acceptable, the flop gate will open to a 5' x 10' double deck vibrating screen. The top deck screen has 2" openings and the bottom deck has ½" openings.

The minus ½" material is fed to #5 conveyor which delivers it to the fine ore storage bin. The plus ½" and the top deck screen oversize is fed to a 32" x 40½" Hammermill. The Hammermill discharge, in the case of open circuit, is delivered to the fine ore storage bin via conveyors 3 and 5. If the circuit is operated on closed circuit, the Hammermill discharge will return to the double deck screen via conveyors 3, 4, and 1. Conveyor #3 is reversible, so that it can be operated on either closed or open circuit.

The stockpile ore is reclaimed through a reclaim hopper on Conveyor number 4.

The fine ore storage bin has a live capacity of 1250 tons. It has a diameter of 35' at the top with a 12' 6" vertical side wall, which tapers at 70° to a 3' by 7' rectangular opening at the bottom. The bin is designed to minimize material hang-ups.

3-2 GRINDING AND THICKENING (See Fig. 2)

The work index for the ore is 8.3 KWH/ton. The grind is 1% +65 mesh.

The grinding is done at a single 8' x 10' ball mill in closed circuit with a 15" cyclone. The cyclone overflow flows by gravity to the 80' thickener. The thickener underflow density is controlled at approximately 50% solid. Approximately 0.1 lbs. per ton settling agent is used in the thickener.

3-3 LEACHING AND HEAT RECOVERY

The thickener underflow is delivered to three preleach pachucas 15½' x 55'. The pachucas have the capacity to condition the pulp for 17 hours at 170° F. with total air consumption between .7 and 1.0 standard cubic feet per minute per ton ore. The pulp is then discharged to two parallel banks of five autoclaves. The autoclaves are 12' diameter x 14'. The leaching temperature is maintained at 250° F. by steam coils, and the pressure at 60 psig. Compressed air is introduced to the autoclaves at the rate of 1.25 standard cubic feet per minute per ton ore. The retention time in the autoclaves is about 7 hours. Leaching conditions are 27-30 gpl Na_2CO_3, 7-9 gpl $NaHCO_3$ and 50% solids.

The overall reaction which covers the dissolution of simple Uranium oxides in carbonate-bicarbonate solutions are:

$$2UO_2 + O_2 = 2UO_3$$

$$UO_3 + Na_2CO_3 + 2NaHCO_3 = Na_4UO_2(CO_3)_3 + H_2O$$

The leached pulp is discharged by means of a level controller through concentric pipe heat exchangers to a primary filter feed holding tank.

After flowing through the heat exchangers mentioned, the temperature of the leached pulp is lowered from 250° F. to 150° F.

3-4 FILTRATION

The removal of the uranium value from the leached pulp is by means of three stages of counter-current filtration. Each filter stage contains four 11½' x 16' string filters.

The feed to each stage of filters is pumped from the holding tank through a pipe-loop. Each filter tub has a level controller which operates a valve on the feed line from the loop.

The first stage filter cake is washed and repulped with hot filtrate from the third stage. The filtrate is sent to Whitco clarifiers. The repulped cake is filtered by the second stage filters, and the filtrate is pumped to the thickener. The cake is washed and repulped with recarbonated barren solution. It is then filtered by the third stage filters. The filter cake is washed and repulped with reclaim water and pumped to tailings.

All repulping solution for the first stage and second stage filters is heated to 160° F. by means of shell and tube type heat exchangers.

The flocculant is diluted before being fed to the specially designed launders so that the flocculant will mix with the filtration pulp before entering the filter tub.

3-5 CLARIFICATION

The primary filtrate is clarified with Whitco vacuum filter leaves. The clarified solution is pumped through the concentric heat exchangers on the discharge of the leaching autoclave. The solution temperature is raised from 120° F. to 170° F. It is then pumped to the reseeding tank of the precipitation section. The slime on the filter leaves is washed off occasionally, and is sent to the preleach thickener.

3-6 PRECIPITATION

Precipitation of uranium is done in a series of seven tanks giving a total retention time of approximately 8 hours. The hot clarified pregnant solution is mixed with recycled yellowcake in the first tank (reseeding tank). The 50% caustic solution is added to the 2nd and 3rd tank to maintain the free NaOH content between 4-6 grams per liter. In this step, the excess bicarbonate is neutralized and enough excess caustic added to obtain the following precipitation reactions:

$$NaHCO_3 + NaOH = Na_2CO_3 + H_2O$$

$$2Na_4UO_2(CO_3)_3 + 6NaOH = Na_2U_2O_7 + 6Na_2CO_3 + 3H_2O$$

The yellowcake slurry from the precipitation circuit flows into a 24' diam. x 10' thickener. Part of the thickener underflow is recycled to the first tank in the precipitation circuit, the balance of the underflow is pumped to the sodium diuranate (SDU) storage tank.

The thickener overflow is pumped through a pressure filter for final clarification and then to the barren storage tank.

3-7 RECARBONATION

The barren solution from precipitation contains sodium carbonate and 4-6 grams per liter caustic. The solution is pumped to a 3' diameter x 68' packed tower where it is contacted with flue gas (CO_2) to regenerate bicarbonate and to convert any sodium diuranate (SDU) present back to uranyl tricarbonate. The regenerated solution is then pumped to the second stage filters for washing and repulping.

Recarbonation reactions are shown by the following equations:

$$2NaOH + CO_2 = Na_2CO_3 + H_2O$$

$$Na_2CO_3 + CO_2 + H_2O = 2NaHCO_3$$

$$6NaHCO_3 + Na_2U_2O_7 = 2Na_4UO_2(CO_3)_3 + 3H_2O$$

3-8 PRECIPITATE RETREATMENT AND DRYING

The sodium diruanate (SDU) slurry from the storage tanks is dewatered by a 28" x 42' centrifuge. The centrate is returned to the SDU thickener. The SDU can be sent directly to the hearth for drying and packaging if contract specifications will allow for a high sodium content. If the contract requirements will not allow a high sodium content then the SDU needs to be refined one more step. This consists of sulphuric acid (H_2SO_4) dissolution, precipitation with hydrogen peroxide, and then stabilization with ammonia (NH_3) at a pH of 3.5. The simplified reactions proceed as follows:

$$UO_2^{++} + SO_4^{--} = UO_2SO_4$$

$$UO_2SO_4 + H_2O_2 = UO_4 \cdot 2H_2O + H_2SO_4$$

The precipitate, uranium peroxide (UPO), is dewatered and washed by two 28" x 42" centrifuges. The centrate is pumped to a pressure filter with the centrate going to tailings as waste. The UPO is stored in a holding tank and pumped by a Moyno pump to a 6' diameter x 6 hearth dryer fired by natural gas. The dried cake is extruded from the hearth through a rotating grizzly to a hammermill and packed in 55 gallon drums for market.

4) ENVIRONMENTAL PROTECTION

An in-plant monitoring system designed to meet and exceed government requirements is in force at the Lisbon Mine.

Sampling of air, water, soil and dust had been started well before start-up to establish a base line for various contaminants. General air samples as well as breathing zone samples are taken to give a good coverage of the whole operation. A sampling grid for soil sampling within 4000 feet of the concentrator has been established. Particulate sampling of discharge stacks as well as routine checks on filter bags is done. A series of ambient air samples up to 3500 feet from the plant are taken each week.

Sampling locations are determined by wind direction and velocity at time of sampling and samples are taken over an 8 hour period.

Regular water samples are being analyzed from several points on the drainage basin from the mine when water is flowing. In addition, monitoring wells have been placed downstream from the tailings dam to intercept any possible seepage. Samples are taken regularly and analyzed for any contamination.